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MILLING METHODS AND COSTS AT THE VERDE CENTRAL CONCENTRATOR, JEROME, ARIZ.

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MILLING METHODS AND COSTS AT THE VERDE CENTRAL CONCENTRATOR, JEROME, ARIZ.[1]

By R. H. Dickson[2] and E. M. Smith[3]

INTRODUCTION

This paper, which describes the milling practice of the Verde Central Mines, Inc., is one of a series on milling methods and costs being prepared by the United States Bureau of Mines.

GENERAL DESCRIPTION

The Verde Central concentrator is located in Walnut Gulch, about 1 mile west of Jerome, Ariz., and on the Jerome-Prescott highway. It is about 1,000 feet distant from the main shaft of the Verde Central mine and treats ore from this mine only. The mill is connected with the main shaft of the mine by a 24-inch gage railway, over which the ore is hauled in 5-cubic yard capacity, rocker, side-dump cars, by a 3-ton Brookville gasoline locomotive.

The concentrator was designed to treat 250 tons of ore per day, but since the installation of a Symons cone crusher and two additional flotation cells it has been handling 400 tons per day.

The concentrates produced are hauled by truck to the Verde Tunnel and Smelter Railway, 2 miles distant, for shipment to the United Verde Extension Smelter at Clemenceau.

Water for milling purposes is obtained from the mine. Power is purchased from the Arizona Power Co. at 66,000 volts, and this voltage is reduced to 440 for use in the concentrator. Two power circuits to the plant have been provided by the Arizona Power Co.; one from Prescott, and the other from Tapco; this provision has reduced to a minimum delays due to lack of power.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6489."

[2] One of the consulting engineers, U. S. Bureau of Mines, and manager, Verde Central Mines, Inc.

[3] One of the consulting engineers, U. S. Bureau of Mines, and mill superintendent, Verde Central Mines, Inc.

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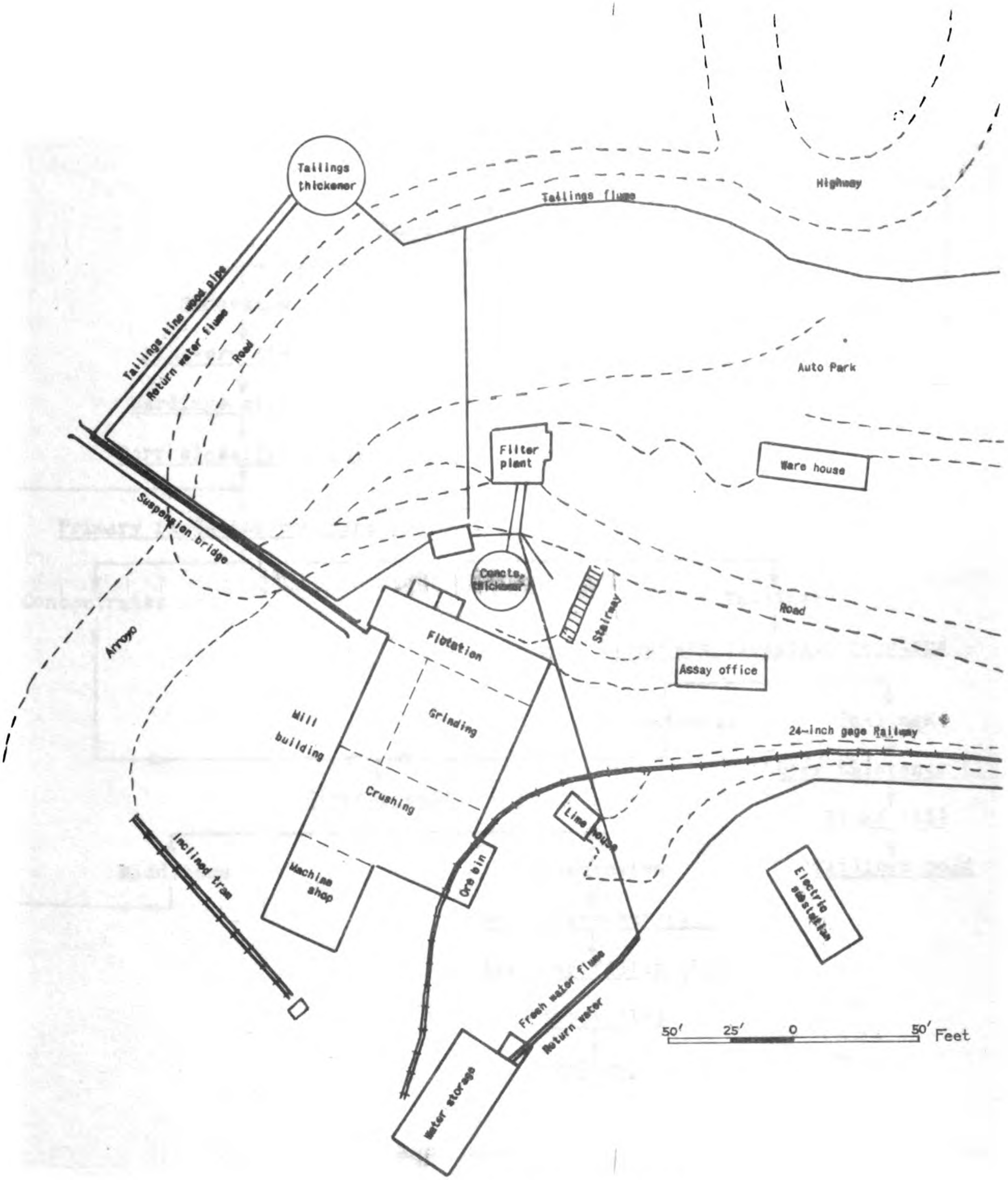

Figure 1.- Ground plan of centrator plant

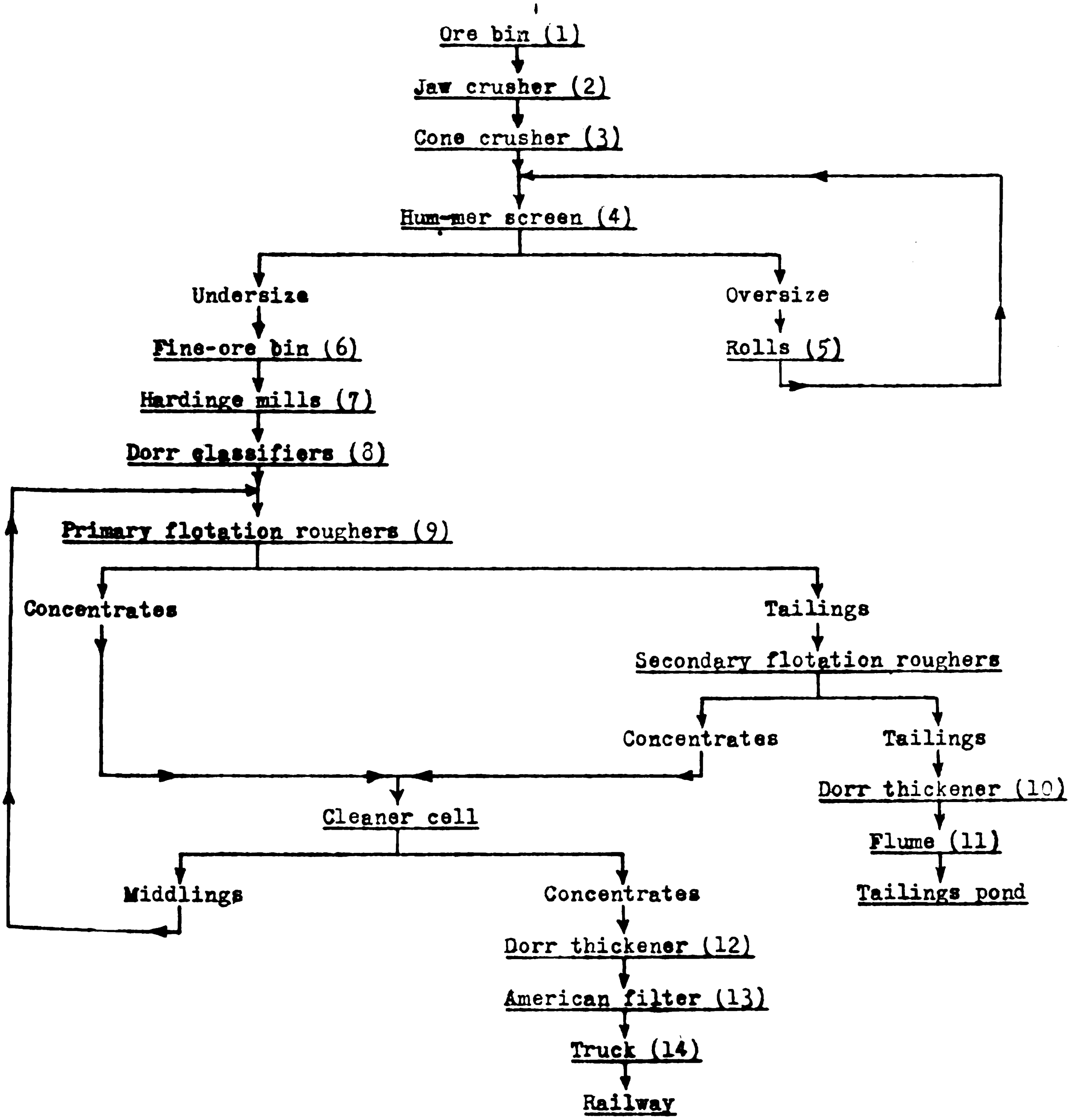

Figure 2.--General flow sheet of the concentrator

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Crushing

Ore is delivered from the coarse-ore bin to a grizzly with 3-inch openings. The oversize is crushed to $2\frac{1}{2}$-inch size by a 12 by 24 inch Allis-Chalmers Blake-type crusher which is equipped with manganese-steel jaws and cheek plates. The crushed product and the undersize of the grizzly join and are conveyed to a No. 3 coarse Symons cone crusher by a 16-inch conveyor belt. A stationary electromagnet is suspended over this conveyor for the removal of tramp iron.

The Symons crusher reduces the 3-inch feed to 5/8-inch size but must be reset twice daily, at 7 a.m. and 7 p.m., in order to maintain the product at this size. The crushing surfaces are of manganese steel; a mantle lasts about 40 days and a bowl liner about 30 days. This machine has a greater capacity than the other machines of the crushing circuit unless the ore contains an unusual amount of wet fine material. On the recommendation of the manufacturer, a 60-hp. slip-ring motor was installed to operate the Symons crusher, but tests show an average load of 35-hp. with a maximum load of 47-hp. when it handles 30 tons of ore per hour. The product of the cone crusher passes by gravity to a bucket elevator which in turn delivers it to a Hum-mer screen.

The Hum-mer screen is 48 by 60 inches in size and has $\frac{1}{4}$ by 1 inch holes. It is installed at a high point in the mill building so that the undersize drops onto a conveyor belt which carries it to the fine-ore storage bin. The oversize falls through a chute and is fed to a pair of 56 by 24 inch Allis-Chalmers Anaconda-type rolls set $\frac{1}{4}$ inch apart. The rolls operate in closed circuit with the Hum-mer screen, the roll product dropping into the same elevator boot as the cone-crusher product. Rolls shells 56 inches in diameter have recently been used and give proportionately longer service as compared to the 54-inch shells. As originally operated, the rolls had a speed of 112 r.p.m., but this has been reduced to 85 r.p.m., and the change has resulted in less corrugating, less power consumption, and greater capacity.

The crushing department has a capacity of 25 to 27 tons per hour. Provision is made so that either the cone crusher or the rolls can be eliminated from the circuit, but operations under these conditions are reduced in efficiency. All machines in the crushing plant are driven by belts.

The total cost of crushing during the period June 1 to September 1, 1930, was $0.346 per ton of ore handled. This cost was distributed approximately as follows:

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Distribution of expense of crushing	Per ton of ore
Operating labor for entire crushing plant	$0.087
Jaw crusher:	
Repair labor	.008
Supplies	.012
Cone crusher:	
Repair labor	.007
Supplies	.048
Rolls:	
Repair labor	.010
Supplies	.061
Miscellaneous[1]	.034
Power, including the power used by conveyors, elevators, Hum-mer screen and dust fan	.079
Total	$0.346

[1] Includes repair work done on conveyors, elevator, pan feeder, and Hum-mer screen.

Maintenance of crushing machinery

The larger pieces of ore produced by the shrinkage-stope method of mining take a heavy toll from the jaw crusher. Fixed and swing jaw manganese-steel plates last four and six weeks, respectively. The pitman cap and the lower half of the pitman are changed and rebabbitted every three months. Roll shells handle 35,000 to 42,000 tons of ore before being discarded. Corrugating of the roll shells is reduced by fleeting one roll daily. Flanging occurs on the shells, but these flanges can be removed by a torch in about one hour and thirty-five minutes.

The cone crusher is lubricated by a force-feed system, the oil pump being direct connected to the drive. The checking of oil is an important item in the operation of this machine; the amount of oil in the reservoir is determined each day and recorded on the daily shift report. A sight-oil feed on the machine is used to check the operation of the oil pump.

The liners of the cone crusher are inspected daily in order to be prepared when a change of liner is required. The changing of a liner is done in four hours by a crew of five men although a smaller crew could do this work if time were not a vital factor. In making this change the belt, feed hopper, and feeder plate must be removed before removal of the bowl casting. The old bowl liner is then detached and a new one bolted and zinced into place. The adjusting threads are cleaned and the machine assembled. A piece of lead is used to test the size of opening and after final adjustments are made the machine is ready for operation. If the mantle is to be replaced, the bowl assembly is removed, after which the shaft head and mantle are removed. A spare shaft and head, with mantle zinced in place, are then put in position and the machine assembled. A 5-ton capacity chain block with a crawl is mounted over and serves the cone crusher.

The fine-ore storage bin is a circular wooden tank 20 feet in diameter by 13 feet high. It is made of Douglas fir with sides of 3-inch staves and bottom of 12 by 12 inch timbers. It is equipped with drum feeders which feed the ore to a conveyor, which in turn delivers it to the scoop boxes of the ball mills. The ore is weighed by a Merrick weightometer enroute to the grinding mills.

Grinding

Grinding is done by two 8-foot by 36-inch Hardinge ball mills arranged in parallel and each operated in closed circuit with a 6 by 26 foot Dorr classifier. The mills are operated at a speed of 15 r.p.m. Cast-iron balls, 3-inch size, made by the Capital Foundry of Phoenix are used, the replacements amounting to 175 balls per day per mill. Experiments with 3-inch and 4-inch forged chrome-steel balls indicated that they were more expensive per ton of ore ground than cast-iron balls. Liners are also of white cast iron, and tests using manganese-steel wedge bars and liners indicated that although the steel liner lasted longer than one made of white iron it was more expensive per ton of ore ground. Liner consumption in terms of pounds of liner consumed per ton of ore ground is not available, but the cost of liner wear over a three months' period amounted to $0.01 per ton of ore ground. The slow operating speed of the grinding mills reduces the wear on liners to a minimum.

The classifiers are considered and operated as a part of the grinding circuit. The rake speeds are 27 strokes per minute. Rakes formerly used were 3 inches deep but these were replaced with others 4 inches deep, with improved results. The classifier sands contain 36 per cent of minus 48-mesh material and the classifier overflow solids contain about 4 per cent of plus 48-mesh sizes.

Flotation

A detailed flow sheet of the flotation circuit is presented in Figure 3. The classifier overflow pulps of the grinding circuits are treated in two primary MacIntosh flotation machines operated in parallel and as roughers. The tailings of the two primary machines are mixed and then distributed to two secondary MacIntosh machines also operated in parallel and as roughers. The tailings of the secondary roughing machines comprise the final waste product of the concentrator. The concentrates produced by primary and secondary machines are delivered to one MacIntosh cleaner cell by a 2-inch Wilfley pump. The cleaner cell produces finished concentrates and middlings, which flow by gravity to the heads of the primary rougher cells. All MacIntosh machines are equipped with 15-foot rotors which revolve at a speed of 18 r.p.m. A No. 3 Roots blower equipped with Timken roller bearings supplies air at 3 to 5 pounds pressure. Blankets used are 3-ply and are purchased from the National Filter Co. A blanket lasts from 30 to 40 days but must be scrubbed every five days.

Reagents used comprise lime, xanthate, and pine oil. The lime is added to the ball-mill scoop boxes by a Denver dry reagent feeder. Titrations are made hourly on primary-cell tailings water in order to control alkalinity which is maintained at 0.20 pound of CaO per ton of water. It has been found that a deficiency of lime in the pulp produces a dead froth and causes high loss of copper in the tailings, whereas an excess of lime makes a very active froth and one that contains a high insoluble content.

Xanthate in the form of potassium ethyl xanthate is normally used but under certain conditions potassium amyl xanthate is used with beneficial results. Xanthate reagents are added to the classifier overflow pulps by a Geary reagent feeder as 10-per cent solutions.

Half of the G. N. S. No. 5 pine oil used is added to the classifier overflow pulps and the other half to the concentrates launders of the cleaner flotation cell for the purpose of breaking down the froth in the thickener. Most of the pine oil added to the concentrates froth returns to the flotation cells with the reclaimed water.

The amounts of reagents used per ton of ore treated follow.

	Pounds
Lime	2.25
Xanthate	0.07
Pine oil	0.05

The primary rougher flotation cells produce concentrates which contain about 15 per cent of copper and tailings which assay about 0.60 per cent of copper. The secondary rougher machines reduce the tailings copper content to between 0.24 and 0.16 per cent.

Screen analyses of flotation heads and tailings for a period of eight days are given in the tabulation which follows.

Screen size mesh	Flotation heads		Flotation tailings	
	Weight per cent	Copper per cent	Weight per cent	Copper per cent
Plus 65	3.5	0.64	5.0	0.37
Minus 65, plus 100..	9.0	0.69	12.5	.41
Minus 100, plus 150.	10.5	1.32	13.0	.15
Minus 150, plus 200.	11.0	2.78	11.0	.12
Minus 200	66.0	4.12	58.0	.23
Composite	100.0	3.24	99.5	.24

The concentrates grade is maintained as nearly as possible to conform with the following analysis:

	Per cent
Copper	20
Iron	29
Insoluble	15

When concentrates of lower insoluble content are produced or a head of higher copper content is treated the copper content of the tailings increases. The ideal grade of concentrates is the one most profitable when tonnage, recovery, freight, and smelting charges are considered.

DEWATERING AND HANDLING CONCENTRATES

The finished flotation concentrates pulp contains about 30 per cent of solids and flows by gravity to a Dorr thickener which is 24 feet in diameter and 8 feet deep. The thickener overflow water passes through a small rectangular tank for final clarification and from there is conveyed to the return water sump. The discharge of the thickener contains from 50 to 70 per cent of solids and flows by gravity to a 6-foot American filter equipped with five disks.

The filter is operated approximately 9 hours per day. When the filter is not operating, the concentrates are stored in the thickener, the rakes being raised or lowered to conform to this method of operating. Vacuum for the filter is maintained by a 12 by 10 inch Sullivan vacuum pump. A barometric standpipe drains the water from the vacuum tank, and this water is returned to the concentrates thickener by a small centrifugal pump. Very fine concentrates accumulate in the tank which handles the discharge water of the barometric standpipe. This material settles over the inlet pipe and is pumped out of the tank intermittently after stirring with a jet of water at high pressure.

One operator attends to the concentrates thickener, the tailings thickener, the filter and the return water pump, and is also available for helping to load the concentrates.

The filter produces about 55 tons of concentrates cake per day, which contains 11 per cent of moisture. The concentrates discharge onto a conveyor belt which is mounted on a car frame served by a track. The track runs over six rectangular bins so that the car can be spotted to discharge the filter cake into any one of the bins. Each bin can hold 5 tons of concentrates and is equipped with an arc gate at the bottom. The contents of two bins constitute a load for the Mack truck which hauls the concentrates to the railroad siding, a distance of 2 miles. The cost of hauling these concentrates, including truck repairs, is $0.25 per ton.

DEWATERING AND DISPOSAL OF TAILINGS

Final tailings from flotation treatment are conveyed to a 30 by 8 foot Dorr thickener through a 5-inch wooden-stave pipe line which is supported on a suspension bridge over Walnut Creek. The pulp fed to the thickener contains 21 per cent and the underflow pulp 35 per cent of solids. Thicker underflow pulps can be produced, but a pulp of 35 per cent solids provides sufficient return water and this consistency is desired for conveying the solids to the tailings pond.

The overflow water from the tailings thickener joins the overflow of the concentrates thickener in a concrete sump and both are returned to the main water-storage tank by a motor-driven pump which is controlled by a float switch.

The thickened tailings pulp is conveyed for a distance of 2 miles to the tailings pond by a launder, 6 by 6 inches inside dimensions, set on a 4-per cent grade. This pond is located within an area where there is a minimum danger of floods washing away the dam. The dam is formed by discharging the pulp at its face and draining the water by means of a well which is located upstream from the dam. One man maintains the face of the dam by shoveling. New trestles for supporting the launder are built from time to time. Water is not returned to the mill from the tailings pond.

SAMPLING AND METALLURGICAL CONTROL

Regular daily samples are taken of flotation heads, concentrates, and tailings. Automatic samplers are used for heads and tailings. The sampler for heads is placed at the junction box which receives the overflows from the two classifiers, and the sampler for tailings is located on the tailings launder just following the secondary rougher flotation machines. The sample of concentrates is taken by hand as the concentrates pulp discharges into the thickener. In addition to the samples noted, daily samples of concentrates are taken by a pipe sampler from the truck, and special samples are taken by hand where and when desired.

All samples are prepared for analysis by the assay office force. A pressure filter, made from a section of 10-inch pipe and using compressed air from the mine, is used for dewatering sample pulps. The cake produced by the filter is dried in about 20 minutes in an electric oven. The dried samples are then broken up by forcing them through an 8-mesh screen, after which they are mixed and reduced to the amounts desired for assay pulps. Tests have indicated that for routine assay work no additional grinding of samples is necessary.

The head samples are assayed for copper and iron, the tailings samples for copper and the concentrates samples for copper, iron, and insoluble. Routine copper analyses of mine and mill samples are made by the permanganate method. The pipe samples of concentrates are used for moisture determinations and the dried residues are used for making composite samples which represent carload shipments. The copper analyses of these composits and also of the control samples received from the smelter are made by a modified iodide method.

The densities of ball-mill discharge and classifier overflow products are determined several times each shift by weighing a known volume of pulp contained in a specific-gravity bottle.

Alkalinity determinations are made, on the water of pulps taken at various places in the mill circuit, by titrating with $\frac{N}{10}$ sulphuric acid solution and using phenolphthalein as the indicator. The best metallurgical results in flotation treatment are obtained when the water in the primary roughers contains 0.2 pound of CaO per ton.

A screen analysis of the final tailings is made each shift. This analysis assists in the operation of the ball mills and classifiers. Screen analyses of other products are made when making special tests.

A Minerals Separation subaeration laboratory flotation machine is used for test work.

WATER SUPPLY

As previously stated, water for mill operations is obtained from the mine. The supply is conserved for future work as much as possible by bulkheading crosscuts and plugging diamond-drill holes that show considerable water. If additional water is required, one or more of these sources is tapped.

All pumping of water is done on the graveyard shift when the mine compressors are not operating, so as to maintain the electric-power demand as uniform as possible over the 24-hour period. Water from the mine flows into a small tank of 6,500 gallons capacity from which water is taken for the glands of the flotation machines. The overflow from this tank passes to the main concrete storage tank which has a capacity of 250,000 gallons. All reclaimed water is pumped directly to this main storage tank.

The concentrator uses about 144,000 gallons of new water per day. The fresh water used is 417 gallons and the reclaimed water 500 gallons per ton of ore milled. As previously stated all reclaimed water is from the Dorr thickeners and the filter, and no water is returned from the tailings pond. There are no soluble salts in the water supply which are deleterious to flotation treatment.

OPERATING DETAILS

The day shift crew comprises 10 men and the other two shifts 6 men each. The distribution of these crews is as follows.

Each of the three shifts is composed of:

- 1 Shift boss
- 1 Crusher feeder
- 1 Crusher operator
- 1 Ball-mill operator
- 1 Flotation operator
- 1 Filter operator

On day shift there are the following in addition to the above:

- 1 Mill superintendent
- 1 Repair man
- 1 Laborer
- 1 Attendent at the tailings dam

The shift boss is held responsible by the mill superintendent for the work of the entire shift. Operators attend to the lubrication of their respective machines and the cleaning of the mill.

The assay office force consists of an assayer and helper. Samples from both mine and mill are handled by these men.

Two roads, one above and the other below the mill, permit the delivery of supplies by trucks. An inclined tramway is available for delivering roll shells, jaw plates and other heavy parts to the grinding section. Grinding balls and lime are dumped into their respective bins from trucks spotted on the upper road. Balls are conveyed by gravity from the bin through an inclined 5-inch pipe line directly to the supply box which is located opposite the discharge ends of the ball mills. Lime, brought into the mill in wheelbarrows, is weighed and fed to the Denver dry-reagent feeder each shift.

All machines are served by overhead hoists and trolleys. The grinding section is equipped with a 21-foot traveling beam which supports a 2 ton capacity Wright high-speed hoist. The rolls have two Wright 8 ton capacity high-speed hoists which are suspended from Wright geared trolleys. The jaw and cone crushers are served by 2 and 5 ton capacity hoists, respectively, which are operated on trolleys with I beams in turn supporting the tracks. The flotation department has an overhead trolley and hoist for handling flotation-machine rotors.

The general machine shop is located adjacent to the mill building. A mechanical department, which serves both mine and mill, takes care of all mill repairs that can not be handled by the operating crew.

Table 1 presents metallurgical data for the months of May and June, 1930.

COSTS

Table 2 gives a summary of concentrator costs for the period June 1, 1929, to August 1, 1930; Table 3 presents detailed concentrator costs for November, 1929.

Table 1.- Metallurgical data for May and June, 1930

Ore treated	tons	21,070
Total operating time	days	58
Ore treated per 24 hours operating time	tons	392
Feed to primary grinding mills	size	minus $\frac{1}{4}$-inch
Screen analysis of flotation feed:		
Plus 48 mesh	per cent	3.5
Plus 65 mesh	do	9.0
Minus 200 mesh	do	51.0
Analyses:		
Copper, total in mill feed	do	2.67
Copper, total in final tailings	do	0.20
Copper in concentrates	do	19.83
Iron in concentrates	do	28.9
Insoluble in concentrates	do	16.1
Moisture in concentrates	do	11.1
Concentration ratio	tons into 1	7.95
Recovery of copper:		
Calculated from assays	per cent	93.3
Based on weights and assays of feed and concentrates	do	93.2
Copper shipped	pounds	1,050,276
New water consumption per ton of ore treated,	gallons	417
Ball consumption per ton of ore	pounds	3.5

Table 2.- Summary of concentrator costs for the period June 1, 1929 to August 1, 1930

Ore milled, tons 139,203
Concentrates produced, tons 16,123

Division of concentrator expense	Power	Labor	Supplies	Total
Crushing mine ore to $\frac{1}{4}$-inch	\$0.073	\$0.143	\$0.161	\$0.38
Grinding	.170	.079	.121	.37
Concentrating	.047	.090	.051	.19
Dewatering and filtering	.005	.042	.007	.05
Tailings disposal	.002	.070	.008	.08
Miscellaneous	.020	.160	.008	.19
Total	\$0.317	\$0.584	\$0.356	\$1.26

Table 3.- Detailed concentrator costs per ton of ore milled in November, 1929

Ore treated, tons 11,384

Division of concentrator expense	Power	Labor	Supplies	Total
Crushing to minus ¼ inch	$0.073	$0.143	$0.161	$0.377
Grinding	.155	.072	.111	.338
Flotation	.037	.078	.039	.154
Dewatering and filtering concentrates.	.005	.042	.007	.054
Thickening and disposal of tailings ..	.002	.040	.040	.082
Reclaimed water	.002	.004	.002	.008
New water	.012	.012	- -	.024
General expense	- -	.018	.009	.027
Repairs to mill building	- -	.003	.003	.006
Supervision, including shiftbosses ...	- -	.083	- -	.083
Assaying	- -	.014	.004	.018
Mill lighting	.005	.002	.001	.008
Ware-house expense	- -	.013	- -	.013
Total	$0.291	$0.524	$0.377	$1.192
Per cent of total	25.1	45.4	29.5	100.0

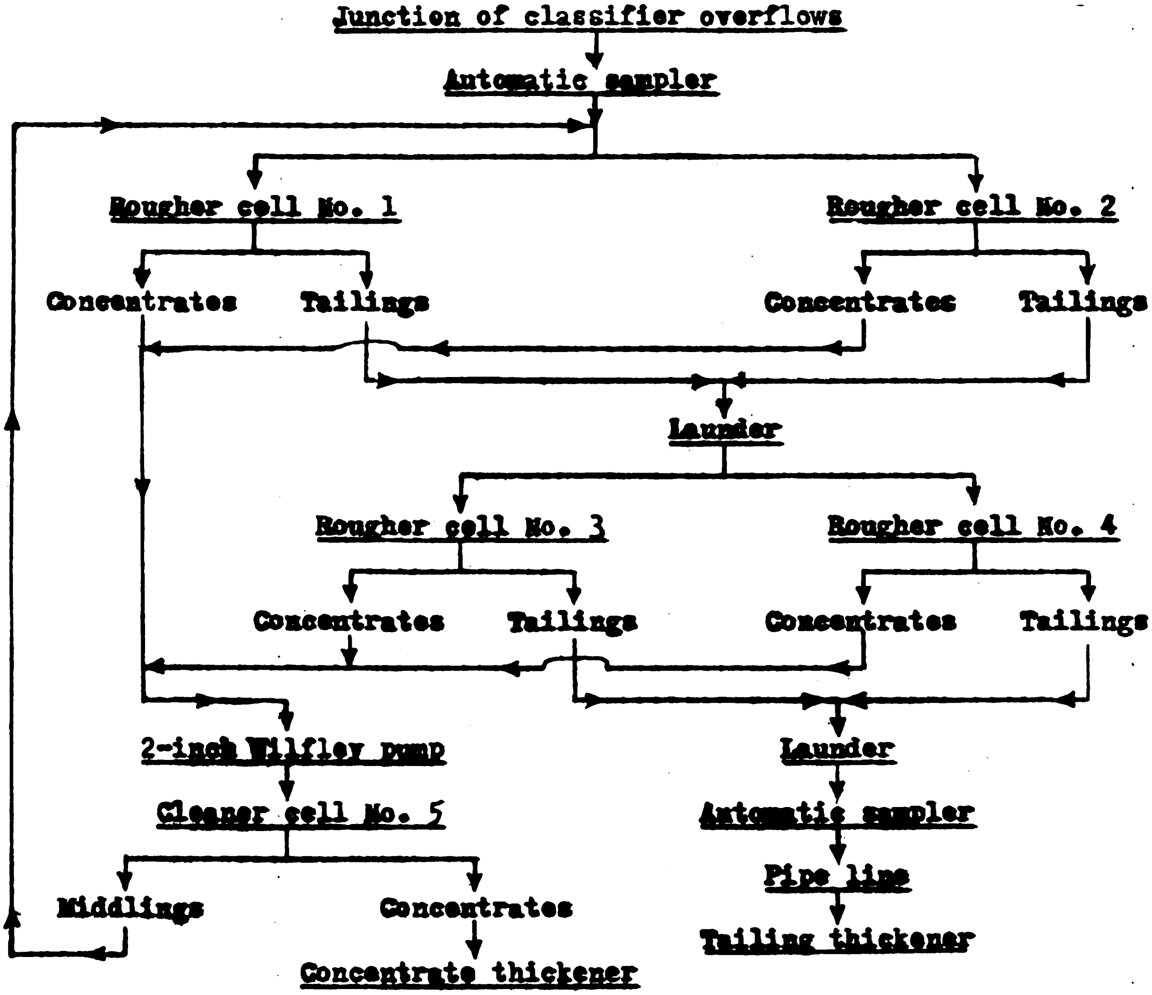

Figure 3.--Detailed flow sheet of the flotation circuit